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In Situ Copper Leaching in the United States: Case Histories of Operations

By Jon K. Ahlness and Michael G. Pojar

UNITED STATES DEPARTMENT OF THE INTERIOR

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UNITED STATES DEPARTMENT OF THE INTERIOR
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As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally owned public lands and natural resources. This includes fostering the wisest use of our land and water resources, protecting our fish and wildlife, preserving the environmental and cultural values of our national parks and historical places, and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to assure that their development is in the best interests of all our people. The Department also has a major responsibility for American Indian reservation communities and for people who live in Island Territories under U.S. administration.

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

°C	degree Celsius
°F	degree Fahrenheit
ft	foot
ft^2	square foot
gal	gallon
gpL	gram per liter
gpm	gallon per minute
gpm/ft^2	gallon per minute per square foot
in	inch
kg	kilogram
kg/d	kilogram per day
kg/mo	kilogram per month
kg/t	kilogram per metric ton
km	kilometer
kPa	kilopascal
L	liter
lb	pound
lb/d	pound per day
lb/ft	pound per foot
lb/mo	pound per month
lb/T	pound per short ton
Lpm	liter per minute
Lpm/m^2	liter per minute per square meter
m	meter
m^2	square meter
m^3	cubic meter
md	millidarcy
mm	millimeter
ms	millisecond
pct	percent
psi	pound per square inch
t	metric ton
T	short ton
yd^3	cubic yard
yr	year

IN SITU COPPER MINING IN THE UNITED STATES: CASE HISTORIES OF OPERATIONS

By Jon K. Ahlness[1] and Michael G. Pojar[1]

ABSTRACT

The copper industry has had a long and interesting history associated with leaching, involving vat, dump, heap, and in situ methods. The Bureau of Mines has also had an interest in copper leaching and has researched the techniques of solution recovery, particularly concentrating on in situ methods. In situ mining is a relatively low cost method and has been proven commercially successful by nine operations. It has been most commonly used for final recovery in old workings at the conclusion of conventional mining operations.

This report brings together information about 10 commercial in situ operations as well as 14 experimental projects. These 24 sites comprise most of the in situ copper mining activities that have taken place in the United States. Background information, geology, ore preparation, solution application, and recovery and processing are provided for each operation. Production data and tables summarizing the engineering statistics for each operation and an extensive in situ mining bibliography are included.

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INTRODUCTION

The purpose of this Bureau of Mines report is to summarize the documented in situ copper leach mining activities that have taken place in the United States. It was prepared to provide a source of engineering and operating data upon which future in situ copper mining activities can be based. Twenty-four different sites are discussed. These are all of the sites that are known to the authors from the extensive in situ copper leach mining bibliography developed at the Bureau's Twin Cities (MN) Research Center (Appendix B).

In situ copper leach mining activities fall into the following categories:

1. Commercial operations with ore body preparation.
2. Commercial operations in old mine workings.
3. Experimental programs.

Ore body preparation includes such activities as blasting, block caving, and hydraulic fracturing (hydrofracing) specifically to fragment the ore body. Operations in old mine workings include those done in open pits, worked out block caved areas, and backfilled stopes where leaching is done as an afterthought to conventional mining. Experimental programs are small-scale tests of the feasibility of a commercial operation. Some of these programs did not progress to the point where leach solutions were applied to the ore.

Each operation (locations shown in figure 1) will be discussed in terms of background information, geology, ore preparation, solution application, recovery and processing, and engineering and production data. Information was collected from published reports, communications with company personnel, and site visits.

In situ mining and the chemistry of leaching copper minerals have been previously discussed (*3, 17, 19, 21, 33*),[2] however, some basics will be covered.

After a suitable leach solution chemistry has been determined, the solutions must be exposed to as much mineralization as possible. This means the ore body must have adequate permeability, either natural through fractures or interconnected pore spaces, or induced by blasting, caving, or hydrofracing. The solution must then be evenly applied to the ore. This can be done with rotating head sprinklers, perforated pipes, lengths of surgical tubing known as Bagdad wigglers, injection holes, or ponds. A method for collecting the pregnant solutions is then needed. This has typically been done with recovery wells, or underground mine workings (raises, ore chutes, and drifts) that direct the solutions to a sump. Pregnant solutions are then treated to remove the values. In the case of copper, treatment is done by precipitation on scrap iron or solvent extraction-electrowinning (SX-EW). In either case, effluent solution from the plant can be recycled back to the leach area.

Because copper-bearing deposits vary considerably in shape, form, and geology, no single method of leaching is universally applicable. Regardless of the preparation method to leach copper successfully the following requirements must be met:

1. Non-acid-consuming host rock.
2. Host rock that will not decrepitate to seal intrarock fractures and block solution flow.
3. Rock sufficiently fractured to permit access of solution to copper minerals.
4. Copper minerals concentrated largely along fractured planes of rock.
5. A solid impervious surface under or surrounding the deposit.
6. Copper minerals that dissolve within required time limits.
7. Ability to recirculate the solution through the ore many times without excessive loss or contamination.
8. Availability of adequate water.

In situ leach mining is a method that can be used to maximize resource recovery at an economical cost.

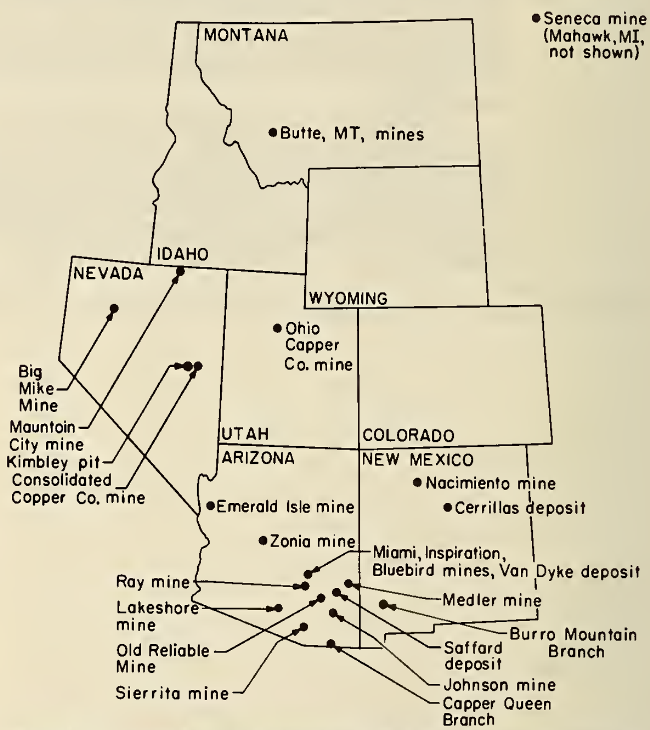

Figure 1.—Copper in situ mining operation and experimental site locations in the United States.

CASE HISTORIES OF OPERATIONS

COMMERCIAL OPERATIONS WITH ORE BODY PREPARATION

This section includes all commercial in situ ventures where ore was prepared for leaching by blasting. Included are the Big Mike Mine, initially an open pit, the Old Reliable Mine, initially an underground operation, and the Zonia mine, initially an open pit-heap leaching operation. The Old Reliable used a "coyote" blast to fragment ore and overburden, and the Big Mike and Zonia used vertical and angled blast holes to fragment ore. All three operations started leaching during the 1972-74 period, but are currently inactive.

Big Mike Mine

The Big Mike Mine is located approximately 30 miles (48 km) south of Winnemucca in Pershing County, NV. Interest in the property dates from the 1900's, however, little exploration was undertaken until the mid-1960's. Ranchers Exploration and Development Corp. obtained the property in 1969 and began a drilling program to confirm earlier exploratory results. The deposit was determined to contain reserves of about 100,000 T (90,720 t) of 10 pct massive copper sulfide

[2]Italicized numbers in parentheses refer to items in the list of references preceding the appendixes.

ore and 700,000 T (635,030 t) of 2 pct mixed oxide-sulfide ore. The massive sulfide mineralization consisted of chalcocite and chalcopyrite, and the mixed ore included cuprite, tenorite, and chalcopyrite. The deposit was approximately 600 by 300 ft (183 by 92 m), and extended to a depth of 300 ft (92 m) from the surface (*38*).

Initial open pit mining began in January of 1970, and by midyear 100,000 T (90,720 t) of the high-grade sulfide ore had been mined along with 300,000 T (272,160 t) of low-grade mixed ore. The high-grade material was shipped to market. The low-grade material was stockpiled, and in late 1971 it was crushed to minus 2 in (51 mm), stacked on an impermeable asphalt pad, and heap leached (*38*).

As a consequence of this method of mining, approximately 475,000 T (430,900 t) of low-grade mixed oxide-sulfide ore was left in the walls and the bottom of the pit. It was decided in mid-1972 to blast this remaining ore into the pit and to leach it in place. Open pit mining had been contemplated, however, it would have required a 6.5:1 stripping ratio and the economic rate of return was considered to be unacceptable (*38*).

On July 10, 1973, 640,000 T (580,600 t) of copper ore and waste rock (average grade of 1.18 pct Cu) was blasted into the open pit using 400,000 lb (181,440 kg) of ammonium nitrate-fuel oil (AN-FO) and waterproof slurry explosives (*38*). One hundred and seventy-five blastholes were drilled around the perimeter of the pit and another 75 in the pit bottom. Three sizes were drilled—5.75-, 9-, and 9.875-in-diam (146-, 229-, and 251-mm). Hole spacing was a 10- by 10-ft (3- by 3-m) pattern for the small holes and a 20- by 23-ft (6- by 7-m) pattern for the large holes. The perimeter holes were up to 300 ft (92 m) deep and angled up to 35°, and the pit bottom holes were up to 100 ft (31 m) deep (*38*). The ore and waste were reduced by blasting to pieces averaging about 9-in diam (230 mm). The shattered ore and waste were then leveled in preparation for leaching. A cross section of the pit before and after blasting is shown in figure 2.

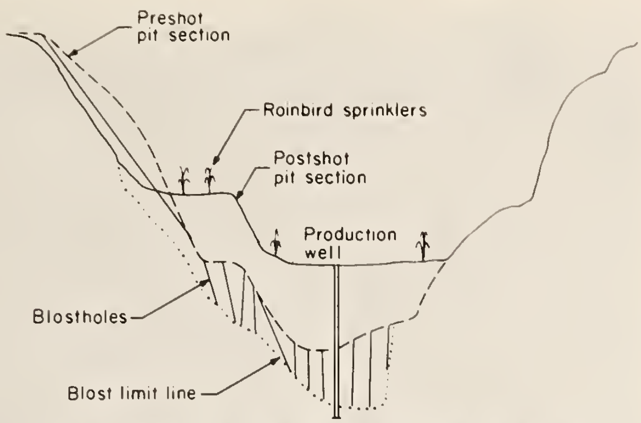

Figure 2.—Big Mike Mine cross section before and after blast.

Leaching was carried out by sprinkling a dilute solution of H_2SO_4 on the ore with Rainbird[3] sprinklers as shown in figures 3 and 4. A water table was present near the bottom of the pit. It was assumed that this coupled with impermeable pit walls would act as a barrier to the percolating solution.

Solution recovery involved one production well drilled through the blasted zone in the pit bottom. A 175-gpm (660 Lpm), 10-stage stainless steel submersible turbine pump was used to pump the pregnant leach solutions to the precipitation plant. Recovery was assisted by the area water table, which was 50 ft (15 m) above the pump. Pregnant leach solution grade averaged 1.0 gpL and did not vary with temperature changes.

[3]Reference to specific equipment does not imply endorsement by the Bureau of Mines.

Figure 3.—Leaching of terrace at Big Mike Mine.

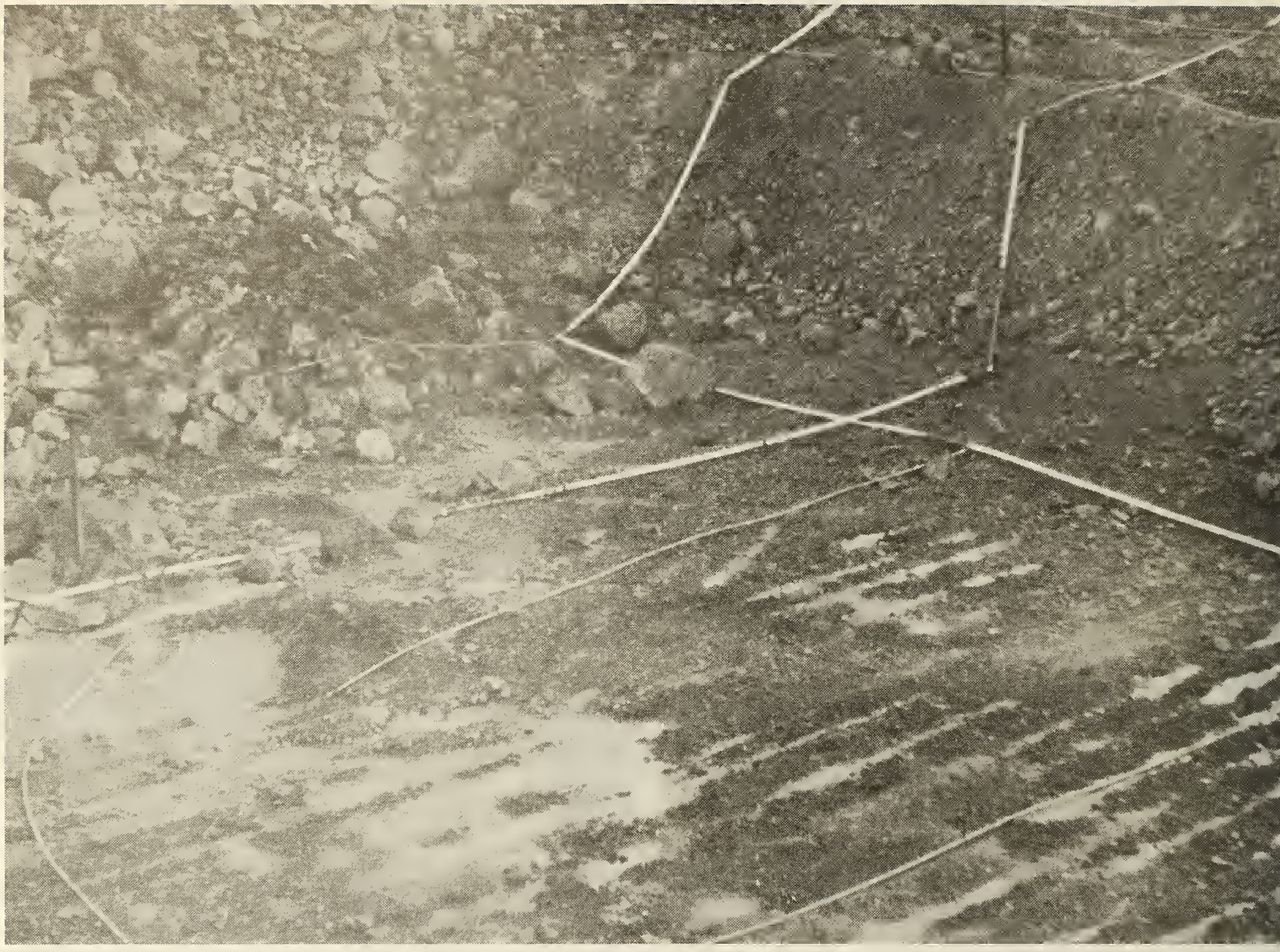

Figure 4.—Pit bottom leaching at Big Mike Mine.

Pregnant solutions from the pit bottom were pumped to a standard iron launder precipitation plant where they were mixed with pregnant heap leach solutions. Cement copper was washed from the concrete cells into a decant area. The copper was then placed on a concrete drying pad where it was turned periodically until the moisture was reduced to an acceptable level, before it was trucked to a railhead and shipped to a Nevada smelter.

The operation was expected originally to produce 5,000 lb/d (2,270 kg/d) of copper for 3 yr. After the initial startup in 1973, however, it was shut down in 1974 when the price of copper fell. It was started again in 1978 and ran until 1979 when the solution grade dropped too low to be economic. There are no current plans for any further leaching at the mine. The operation was economically very successful; development costs were recovered in 6 months. Total cement copper production from both heap and in situ was 7 million lb (3,175,000 kg) which represented a 25-pct recovery. Operating data are summarized in table A-1.

Old Reliable Mine

The Old Reliable Mine is located 40 miles (65 km) northeast of Tucson, AZ, in the Copper Creek area of the Galiuro Mountains. The site is located on ground that was first claimed for its mineral value during the Civil War. Old Reliable was mined sporadically as an underground operation from 1890 to 1919. During the period it yielded approximately 30,000 T (27,200 t) of ore (*18*). Except for one additional short time span between 1953 and 1954, the deposit had gone unmined for half a century. The amount of ore contained in the deposit made further conventional mining economically unfeasible. In 1970 the deposit came under control of Ranchers Exploration and Development Corp.

The ore body is a near vertical breccia pipe (fig. 5). The host rock is andesite, a lava that is the principal rock in the Glory Hole Volcanic Formation. Molten granite intruded on the area of the deposit perhaps 68 million years ago and cracked the volcanic rock. Copper-bearing solutions flowed into the cracks and voids in the brecciated lava. Ground water later dissolved and redeposited the copper deeper in the ore body in concentrated form. The deposit extends from the surface to a depth of 500 ft (152 m). That portion near the surface is low-grade, containing less than 0.4 pct copper. Mineralization increases with depth, however, reaching an average grade of about 2 pct Cu near the bottom of the ore body. High-grade areas within the deposit sometimes reach 9 to 10 pct (*11*). The deposit is sized at 4 million T (3.6 X 10^6 t) of ore averaging 0.84 pct Cu with a 70:30 sulfide-oxide ratio. The most abundant copper minerals are chalcocite, malachite, chalcopyrite, and chrysocolla.

Ranchers Exploration and Development Corp. decided that a combination of blasting and leaching could be used to recover copper values. After a thorough investigation of the ore body, E. I. du Pont de Nemours & Co. recommended a "coyote" type blast. In contrast to the conventional method of drilling vertical blastholes, a coyote shot uses a network of tunnels and crosscuts driven into the deposit.

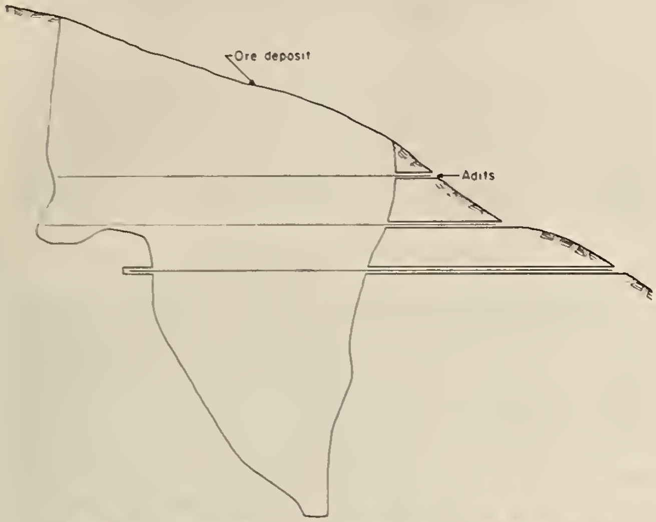

Figure 5.—Cross section through Old Reliable deposit.

The Old Reliable contained 4,000 ft (1,220 m) of tunnels and crosscuts from previous mining, and an additional 4,000 ft (1,220 m) of tunneling was driven for the blast. It was calculated that 4 million lb (1,814,370 kg) of explosives was required for the blast. Eighty thousand 50-lb (23-kg) bags of AN-FO were placed in the tunnels and crosscuts. A wired system of blasting caps, detonating cord, and high-explosive primers was connected to the detonation station in a bunker about a mile from the blast site. The explosive charges were confined by blowing sand into the tunnels and crosscuts and by placing bags of sand for stemming. On March 9, 1972, the shot was fired. Breakage of the ore was considered excellent with pieces averaging approximately 8.5-in diam (216 mm) (*32*). The blast fractured a total of 5 million T (4.5 X 10^6 t) of material averaging 0.78 pct Cu.

Terracing of the fragmented deposit with a bulldozer was completed by May 30, 1972. Figure 6 shows the blasted hillside. A total of 50 million gal (18.9 X 10^7 L) of leach solution was required to saturate the fractured ore body. To provide water to soak the leach area, a 17.5-in-diam (445-mm) well was drilled at a location about 6 miles (9.66 km) from the plant site. The water was acidified with H_2SO_4 at the plant in a barren-solution pond before it was applied to the ore.

A sprinkler system was developed for applying the solution. Two distribution systems existed, each comprised of one 6-in-diam (152-mm) main transmission line and several 2-in-diam (51-mm) distribution lines. Rotating sprinkler heads were spaced at 40-ft (12-m) intervals along the distribution lines. The sprinklers were plastic bodied with stainless steel pins and rings. The front and rear spray nozzles covered a circular area with a radius of 60 ft (18 m) (*18*).

By September, solution was being sprayed on the fragmented ore at a rate of 400 gpm (1,514 Lpm), and production of copper began in January 1973. At full capacity, solution was fed at 1,000 gpm (3,785 Lpm) to the leaching sites to provide 800 gpm (3,028 Lpm) of pregnant liquor for the plant; the difference was lost to evaporation. Design production capacity of the operation was approximately 20,000 lb/d (9,072 kg/d) of copper. Production life was expected to be roughly 5 yr.

Pregnant leach liquors flowed from the base of the rubblized hillside just above the water table into a catchment basin. From there the leach liquors flowed first to a retention pond, then to the cementation launders where the pregnant copper solution was precipitated onto incinerated scrap iron. Six concrete, false-bottomed launders, 8 by 24 ft (2.44 by 7.32 m) were used. Copper precipitated on the iron, in 98-pct conversion, was sluiced off once a day by high-velocity water streams. The settled out copper was periodically removed

Figure 6.—Old Reliable blasted hillside.

onto a solar drying pad. The cement copper, approximately 80 pct Cu on a moisture-free basis, with the balance comprised largely of oxygen, iron, and silica, was shipped to smelters for refining.

The Old Reliable was leached for 2 yr until September 1974, with recovery estimated at 10 pct the first year and 6 pct the second. At that point the dwindling solution grade (0.8 gpL) and a falling copper price dictated shutdown. In August 1979 leaching of the site resumed. Six injection holes supplied an acid leach solution at a rate of 300 to 350 gpm (1,135 to 1,325 Lpm). A similar amount of leach solution was distributed through the sprinklers. Application of leach solution was discontinued again in September 1980, and cement copper production ceased in January 1981, when Ranchers Exploration could not extend the property lease. At the time of shutdown the solution grade was 0.4 gpL Cu.

Total copper recovery during the life of the in situ operation was 13 million lb (5,896,700 kg). This represented only 19 pct of the total copper, but 64 pct of the oxide. A 6-pct copper recovery was needed to break even and was achieved during the first 6 months of operation. Operating data are shown in table A-1.

Zonia Mine

The Zonia mine is located approximately 100 miles (160 km) northwest of Phoenix near Kirkland, AZ. The Zonia mine has been in existence since 1877, but was not operated successfully until it was purchased by the McAlester Fuel Co. in 1966. From 1966 to 1972 the operation consisted of an open pit and heap leach. In 1972 it was decided that the remaining ore reserves would be blasted and leached in situ (*31*).

The deposit occurred in a host rock that was a Precambrian shale and andesite mixture. The major copper-bearing mineral was chrysocolla, which was disseminated throughout the ore body. Minor amounts of azurite, malachite, and tenorite were also present. The ore grade averaged less than 0.3 pct Cu.

Three blasts were detonated between April 1973 and May 1974 to fragment the ore for leaching. Holes for the first blast were drilled to an average depth of 170 ft (52 m). Hole spacings ranged from 11 to 15 ft (3.4 to 4.6 m) depending on hole depth. Table 1 summarizes the blasting data.

Dilute H_2SO_4 was applied at the rate of 0.0017 to 0.0025 gpm/ft^2 (0.07 to 0.10 Lpm/m^2). Application methods included sprinklers and perforated pipes. Leach solutions collected at the base of the ore body and were pumped to the surface through a recovery well. The water table was used as an aid for leach solution collection. Pregnant solutions were processed in a precipitation plant that had a maximum production capacity of 5,000 lb (2,270 kg) of copper per day. In the 2 yr of operation, approximately 10 pct of the copper from the broken ore was recovered.

The Zonia closed in early 1975 because of the low price of copper and a lack of a market for the product, cement copper. The McAlester Fuel Co. was later purchased by another energy fuels company, and the mine was put up for sale. Reserves containing 0.3 pct Cu still remain. Table A-1 summarizes the in situ mining operation data.

Table 1.—Zonia mine blasting data

Date	9-in-diam blastholes		Explosive		Ore blasted, 10^3 T
	Number	Depth, ft	Type	Amount, 10^3 lb	
Apr. 1973 ...	1,710	100-240	AN-FO ...	4,140	4,000
Mar. 1974 ...	NA	NA	AN-FO, slurry.	873	1,050
May 1974 ...	754	110-212	...do ...	1,544	1,780

NA Not available.

COMMERCIAL OPERATIONS IN OLD MINE WORKINGS

In situ mining in old mine workings is discussed in this section. No ore preparation was done; leaching was an afterthought to conventional mining. Of the seven operations in this category, five involved the leaching of old block caved workings, one involved backfilled stopes, and the other, an open pit with underground workings; two are currently active (Copper Queen Branch and Miami mine).

Burro Mountain Branch

The Burro Mountain Branch of Phelps Dodge Corp. was located 10 miles (16 km) south of Silver City, NM, at the present site of Tyrone. Phelps Dodge acquired the property in 1909 and mined 2 million T (1.8 X 10^6 t) of ore by underground methods before the mine closed in 1921 (*30*). The principal copper mineral was chalcocite.

In May 1941, in situ mining of old low-grade caved areas began. Water was percolated through the caved material, collected underground, and the pregnant solution processed in a precipitation plant (*34*). The plant and leaching operations were expanded in 1942 (*30*). Leaching was interrupted by strikes in 1946 and 1948 and was finally discontinued in 1949 (*30*). The area is now part of the Tyrone open pit. Operating data are shown in table A-1.

Butte, MT, Mines

In situ copper mining has been done in the old backfilled stopes of the Anaconda Minerals Co. underground mines near Butte, MT, since the 1930's. The practice evolved from the work of the "fire fill" crew. This crew drilled holes into old stopes and flooded them to extinguish fires. They found that the water disolved significant amounts of copper. In situ leaching was eventually done in three of the six major underground copper mines in the area (Leonard, Mt. Con, and Steward). In 1964, all in situ leaching activities in the area terminated with the introduction of dump leaching, which produced higher grade solutions and accounted for the entire capacity of the solution processing plant. The last of the underground mines closed in 1981, and the Berkley Pit closed in May 1982 (*20*).

The host rock for the ore is a quartz monzonite. Principal copper minerals are chalcopyrite and chalcocite with minor amounts of bornite, azurite, and malachite. The ore occurs in veins that typically dip at 60°. The underground mine workings extend to 5,000 ft (1,525 m) in depth, but most of the leaching was done in the upper 3,000 ft (900 m). Stopes were typically 100 to 150 ft (30 to 46 m) high and 75 ft (23 m) in strike length on both sides of a center service raise. They were backfilled with mine waste that typically graded 0.8 pct Cu, but occasionally as high as 2.0 pct.

Two leaching methods were used, flooding and trickle leaching. Flooding produced the most favorable results. Bulkheads were built to isolate stopes, which were then flooded with water through old mining access points. The stope was filled to the top, drained immediately and allowed to sit for at least 2 months before being flooded again. During the rest period, compressed air was pumped into the stopes to increase oxidation of the material.

Trickle leaching was done by drilling injection holes into the stopes from the hanging wall. The holes were positioned to allow the leach solution to contact the greatest amount of ore before it flowed down along the footwall. An injection rate of 3 gpm (11 Lpm) per hole was used. Higher rates caused channeling. Leach and rest cycles ranged from alternate months to continuous leaching until the grade dropped to an unacceptable level (usually 2 to 3 months) before resting. Pregnant solutions from each level area were monitored, with the arbitrary cutoff depending on grades from several locations.

The first leach solution used at Butte was surface water. Eventually water pumped from the mine for dewatering purposes was used. This water was more effective because its ferric sulfate content aided the leaching process. Total flow rate to the stopes never exceeded 700 gpm (2,650 Lpm). Small amounts of H_2SO_4 were added underground to lower the solution pH to 2.1 from 2.7, which prevented iron salt precipitation. The total number of stopes under leach at one time depended on the other operations in the mine. Most leaching was done when the mine was on a reduced production schedule so the leaching would cause only minimum interference.

Pregnant leach solutions mixed with other mine water in the mine sump. This mixture was pumped at a rate of 5,000 gpm (19,000 Lpm) to the precipitation plant on the surface. The mixed solution contained 0.50 to 0.75 gpL Cu. Operating data are shown in table A-1.

There is still substantial leaching potential in the old stopes, but conventional underground mining would have to be justified, because, while in situ mining more than paid for mine dewatering, it was not economic on its own.

Copper Queen Branch

The Copper Queen Branch of Phelps Dodge Corp. is located at Bisbee, AZ. The area has a copper mining history dating back to the 1880's when production from the mine began. Phelps Dodge bought the Copper Queen in 1885 and operated the underground mine relatively uninterrupted until June of 1975 when low copper prices and high labor costs caused it to be shut down. Phelps Dodge began production from the Lavender Pit in 1954 and operated it for 20 yr until its closure in December of 1974 (*29*). In situ mining of the Lavender Pit, with the solution collected in the underground workings of the Copper Queen, began in 1975 to supplement the dump leaching that had been done since the late 1950's. In situ mining was an economical way to recover more copper from the pit. Both the dump and in situ leaching operations are currently active.

The host rock for the ore in the Lavender Pit is the Sacramento quartz porphyry stock. The principal copper mineral is chalcocite, with minor amounts of azurite and malachite. No ore preparation was done in the pit before leach solutions were applied.

The water used for leaching in the Lavender Pit is pumped from the underground mines. No acid is added because it is generated from the oxidation of pyrite in the ore. The water is sprayed onto the pit walls with approximately 70 sprinklers. Each sprinkler covers a radius of about 35 ft (11 m), but radius of coverage varies slightly with the depth of the sprinkler in the pit. The sprinklers have no pattern or set spacing, but are placed on exposed ore where access is available to personnel. All the accessible areas are presently being leached, and the cost to open more areas is prohibitive. No leach-rest cycle is followed.

Solutions run down the walls and collect at the pit bottom (fig. 7), 950 ft (290 m) below the surface. Two holes drilled in the pit bottom provide passage for the copper leach solutions to the underground workings. The solutions then migrate down to the 1,800-ft (549-m) level where they collect and are pumped to the surface through the Junction shaft. They are mixed with pregnant solutions from the No. 7 dump before treatment in a precipitation plant. The barren solution from the plant is recirculated to the No. 7 dump.

Figure 7.—Solutions collecting at bottom of open pit at Copper Queen Branch.

In situ mining has also been done in the nearby Campbell underground mine. The 1,400-ft (427-m) level was dammed and flooded. Solutions percolated down to the 2,100-ft (645-m) level and were pumped to the solution collection area in the Junction Shaft before being pumped to the surface.

The copper grade of the mixed solution fed to the cementation plant is 0.6 gpL. Total copper production capacity is about 24,000 lb/d (10,896 kg/d) with the in situ portion contributing about 25 pct. Labor requirements for the operation are small. Two individuals can handle the pit's plumbing and only minimal maintenance is needed in the underground workings. Sprinkler plugging is not a problem, because none of the barren solution is recirculated back to the pit. The leaching is planned to continue as long as the pregnant solutions grade remains satisfactory. Operating data for the in situ mining program are shown in table A-1.

Inspiration Mine

The Inspiration mine is located between the towns of Miami and Globe, AZ, about 75 miles (120 km) east of Phoenix. The mine, owned by Inspiration Consolidated Copper Co., was originally a block caving operation but is presently active as an open pit. Dump leaching began in 1950 on waste material that was stripped from the pit. In situ mining of portions of the old, block-caved workings was done from 1967 to 1974.

The host rock for the ore is a granite schist. The principal copper minerals are azurite, malachite, and chrysocolla with minor amounts of chalcopyrite, chalcocite, cuprite, and covellite. The ore grade in the block-caved workings at the time of leaching was below 0.5 pct Cu. The leach area was above the water table.

Preliminary work began around 1965 with the drilling of injection holes. Approximately 30 holes were drilled in a 3- to 4-acre (12,140 to 16,187-m^2) area and were cased with perforated plastic pipe. This area was near Inspiration's property boundary and could not be included in the open pit operation without extending the pit onto the neighboring property. This left in situ leaching as the only economic alternative for recovering more copper from the caved area.

Leach solution was effluent solution from the precipitation plant with 6 to 10 gpL H_2SO_4 added. Solution injection started in 1967 and ranged from 5 to 20 gpm (19 to 76 Lpm) per hole. There was no specific timetable for leach-rest cycles. When the grade from an area dropped below the cutoff level, the area was rested 3 to 5 months. The average pregnant solution grade dropped gradually during the operation.

Solution application was originally handled by four persons working 8-hour shifts, 5 days per week. Their responsibilities were to check solution flows to the weir boxes and to keep acid tanks full. The crew size was eventually cut to one.

The pregnant solutions migrated down through the caved ore and collected in drifts at the 850-ft (260-m) level. They were then pumped up the shaft to the precipitation plant. The plant consumed 200 T (180 t) of scrap iron each month. Copper production averaged 1.9 million lb (8.6 X 10^5 kg) per year during the operation.

Leach solution was no longer applied after 1974 for the following two reasons:

1. The automatic pump malfunctioned and flooded the pump station.
2. An agreement was made with the neighboring property owners to allow Inspiration to extend its open pit onto their land. This allowed the area to be mined as an open pit.

The operation had no major problems. Iron salt buildup was controlled by adjusting the leach solution pH. Solution channeling was handled by shooting small explosive charges in the wells to "shake up" the material around the wells to change flow patterns.

Although in situ mining is not now taking place at Inspiration, dump leaching still is. New material is continuously being added to the leach dumps from the open pit stripping, and a new SX-EW plant was built in 1979. Operating data for the in situ mining program are summarized in table A-1.

Miami Mine

The Miami mine is located near Miami, AZ, about 70 miles (112 km) east of Phoenix; production from it began in 1910. The original ore body averaged 2.0 to 2.5 pct Cu and was mined by sublevel caving and shrinkage stoping methods; however, the principal mining method used for the large quantities of lower grade ore was block caving (*13*). By 1928, 100 million T (90.7 X 10^6 t) of ore graded at 0.88 pct sulfide copper had been delineated for block caving (*23*). All conventional mining ended in June of 1959 after 152.4 million T (138.3 X 10^6 t) of ore had been mined and the ore body had been exhausted. In situ mining of the caved stopes began on a small scale in a worked out portion of the mine in January of 1942, with full-scale leaching beginning when the underground mine was closed in 1959 (*13*).

The host rock for the deposit is the Precambrian Pinal Schist, which has been intruded by the Schultz Granite porphyry, and is partially covered by the Gila Conglomerate. The area is highly faulted and fractured. The principal copper mineral is chalcocite with minor amounts of chalcopyrite, bornite, covellite, malachite, azurite, chrysocolla, cuprite, and native copper. The mineralization occurs in seams, veinlets, and disseminated particles. Reoxidation of the enriched sulfides along two major faults produced some mixed oxide-sulfide ore (*13*).

Block caving of the ore body was done on a checkerboard pattern with blocks being 150 by 150 ft (46 by 46 m) and 150 by 300 ft (46 by 92 m). The caving activity resulted in surface subsidence over a 5-million-ft^2 (4.6 X 10^5-m^2) area creating a "glory hole" 600 ft (183 m) deep. The resulting material left to be leached is 600 ft (183 m) of ore and waste, with the prime leaching target being the bottom 150 ft (46 m). The overlying material only averaged 0.03 pct copper (*13*).

Besides leaching the leftover caved material, two attempts were made to prepare ore specifically for leaching. In 1954, a low-grade portion of the ore body was block caved with just enough draw to break the ore to the surface (*9*), and in 1969 another low-grade portion was mined as an open pit with the ore placed in the glory hole. This involved 1.3 million T (1.2 X 10^6 t) of material grading 0.78 pct. It created a leaching pad area of 200,000 ft^2 (18,580 m^2) that was 135 ft (41 m) thick (*13*).

The first attempt at leaching in 1942 was done with water, but it proved unsuccessful because not enough pyrite was present in the ore to generate the needed acid. A 6.0-gpL H_2SO_4 solution was then tried successfully. Solution has been applied with sprays, ponds, and injection holes, but only sprays and holes are currently used. Ponding was discontinued because it applied the solution at too fast a rate and because of the leach area's unevenness.

Sprays are created with perforated plastic pipes, which are 2-in diam (51-mm) and are punched with an ice pick every 6 to 8 ft (1.8 to 2.4 m). The pipes are placed on terraced areas of the glory hole. These areas have to be reworked periodically when the solution percolation slows or the grade drops off. Reworking is done by bulldozing about 3 ft (1 m) of material off the surface and over the edge.

Injection holes are 6-in diam (152-mm), average 195 ft (60m) deep, and are cased with 2-in-diam (51-mm) PVC pipe, perforated in the bottom portion. A 1-in-diam (25-mm) hose carries the solution from the plastic distribution lines to the casing.

Flows to different leach areas are controlled by weir boxes where acid is added to bring the acid concentration level up to 5.0 gpL. Both sprays and injection holes are fed from an individual weir and, because flow rate is measured only at the weir, the flow into individual holes is undetermined.

Five hundred holes were in use in June 1981 and more were being drilled. Some of these have been used for years and still accept as much solution as is normally injected, while the flow to others has to be reduced to a trickle to prevent overflow. The maximum area of influence from an injection hole is assumed to be a circle 50 ft (15 m) in diameter.

Leach solution percolates down from the surface through the caved ore and then through raises and drawpoints until it reaches the 1,000-ft (305-m) level, which was the final haulage level. This percolation usually takes 3 to 4 weeks from the time solutions are applied at the surface and lasts 2 weeks after application stops. Two drifts on the collection level run under the ore body. Dams in these drifts contain the pregnant leach solutions. Pipelines [16-in diam (406-mm)] carry solutions from the dams to the sump which is 120 ft (37 m) deep. Automatic shutoff valves prevent solution in the sump from rising too high and flooding the pump station. There are 41 underground sampling points for monitoring the leach solution grades. Leach solutions from the mixed oxide-sulfide ore areas have higher grades than those from sulfide areas (*12*).

Solutions are pumped through 12-in-diam (305-mm) stainless steel pipe to the SX-EW plant on the surface by three 1,000-gpm (3,785-Lpm) stainless steel submersible pumps, with a fourth pump as a backup. The pumps are placed in casings in the sump. The SX-EW plant was built in 1976. Previous to this, a precipitation plant designed to produce 50,000 lb/d (22,700 kg/d) of cement copper was used. The SX-EW plant was designed to handle 3,000 gpm (11,350 Lpm) of pregnant leach solution, and to produce 30,000 to 35,000 lb (13,600 to 15,900 kg) of cathode copper per day.

Solutions grades from the initial leaching were 4.2 gpL Cu. In 1963, the time when there were no more new areas to leach, the grade had dropped to about 2.0 gpL and by June of 1981 the pregnant solution grade was 0.85 gpL. Copper recovery by solvent extraction is approximately 90 pct. Temperature of the pregnant leach solution to the SX-EW plant is a constant 69° to 71° F (20° to 22° C), which is ideal for processing. Total copper production through 1981 by in situ mining at Miami was about 275 million lb (125 X 10^6 kg).

The operation currently employs approximately 40 people including management, maintenance, and a 10-person underground crew that keeps the old workings open and collects samples. The life expectancy of the in situ leaching operation is uncertain but to this point has been very successful and profitable. A schematic of the operation is shown in figure 8 and a summary of operating data is given in table A-1.

Ohio Copper Co. Mine

The Ohio Copper Co. mine was located in Bingham Canyon, UT, about 20 miles (32 km) southwest of Salt Lake City. In situ copper leaching took place in the 1920's after

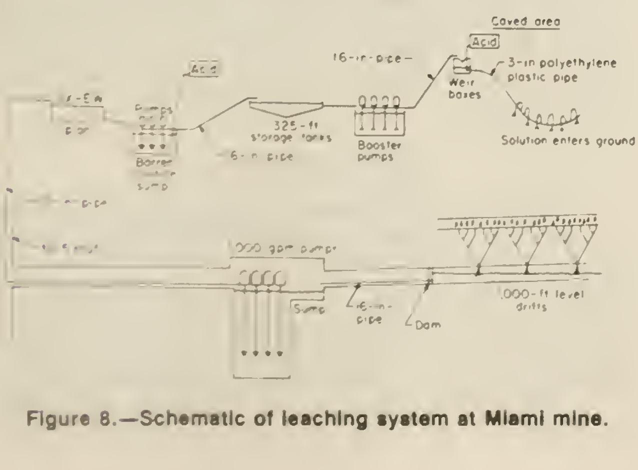

Figure 8.—Schematic of leaching system at Miami mine.

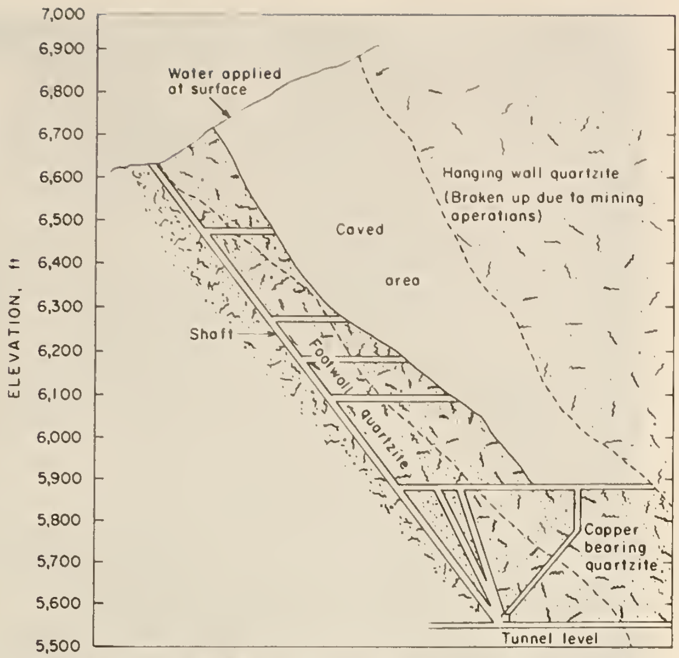

Figure 9.—Cross section through Ohio Copper Co. mine.

the block caving operation shut down when the veins on which mining operations originally depended were more or less completely worked out (*1*).

The host rock for the ore body was quartzite and quartz monzonite. The principal copper mineral was chalcocite which was disseminated throughout. Azurite and malachite were present in minor amounts. The average grade of the estimated 38 million T (34.5 X 10^6 t) in the caved area was 0.3 pct Cu (*40*). Pyrite was also present. The caved zone was in the shape of a truncated cone with the 1,200-ft (366-m) axis 40° off the vertical (fig. 9). The upper dimensions of the cone were 1,400 by 600 ft (427 by 183 m). The bottom area was about one-fourth of the top (*1*).

In August of 1922, 250 gpm (946 Lpm) of water from Bingham Canyon Creek was applied to the surface of a sump over the caved area. The solutions that collected in the underground workings contained 4.5 gpL Cu. With the success of this program, solutions were being applied at a rate of 1,200 to 1,400 gpm (4,542 to 5,300 Lpm) by November of 1923 when mine water was used to supplement the creek water (*1*). Leach solutions were distributed by a 150-ft (46-m) long launder built of three 2- by 12-in (51- by 305-mm) planks with 2-in (51-mm) holes in the sides. The launder was moved on a regular pattern. Later, cascading was attempted to try to introduce oxygen into solution (*6*).

The solutions percolated through the broken material and then through ore chutes to a tunnel 700 ft (214 m) below the bottom of the caved area. The pregnant solutions were processed in a precipitation plant in the tunnel. Copper production was 600,000 lb/mo (272,400 kg/mo), and the plant recovered 97 pct of the copper from solution. Operating data are shown in table A-1.

Ray Mine

The Ray mine is near the town of Ray, AZ, about 70 miles (112 km) southeast of Phoenix. The host rock for the ore body is schist porphyry. The principal copper mineral in the ore mined by block caving was chalcocite; it averaged 1.0 pct Cu and was the result of secondary enrichment. Pyrite content was very high. Above the ore was 125 ft (38 m) of low-grade material averaging 0.6 pct copper, and 50 ft (15 m) of barren capping that ran to the surface (*37*).

The mine closed in 1933 and did not reopen until 1937. This allowed the broken ore that remained in the block cave to oxidize to the point where it was difficult to mill (*12*). Since no further mining operations were planned on lower levels, it was decided to leach the remaining material in situ. Drainage drifts were driven, concrete dams were installed to prevent the solution from flowing into working areas, and a concrete ditch with a 500-gpm (1,900-Lpm) capacity was built to carry pregnant solutions to the pumping station.

Water was first applied to the caved surface on January 20, 1937, using rotary head sprinklers. The leach area covered about 10 acres (40,500 m²) although the entire area was not leached at one time. Pregnant solution grades would decrease with time when the sprinklers were left in one location. It was found, however, that by moving the sprinklers, letting the area rest, and then releaching, the solution grades would return to higher levels. The standard pattern would be to leach an area until the solution grade dropped to 4.0 gpL and then rest the area for 2 months. When solutions were reapplied, the grades would return to 10.0 gpL (*12*).

After 6 months of leaching, substantial settlement of the caved area destroyed the solution distribution pipeline system. Leach solution was then only applied to the edges of the caved area with a few hose lines run into the cave.

Pregnant solutions collected underground and were pumped to the surface through an 8-in-diam (200-mm), lead-lined pipe. Processing was done by precipitation on scrap iron in a plant designed to handle 500 gpm (1,900 Lpm). The plant was operated by six people during the day shift only. All tailing solution was discarded. In the first 18 months of operation, 10 million lb (4.5 X 10⁶ kg) of copper was produced from pregnant solution with an average grade of 9.23 gpL (*36*). Operating data are shown in table A-1. The in situ leach area is now part of the Ray open pit, where present activities are open pit mining and dump leaching.

EXPERIMENTAL PROGRAMS

Experimental leaching programs and fragmentation tests are discussed in this section. Such programs are typically small scale and are used to determine the feasibility of a commercial operation. Fragmentation experiments were done at several sites and did not involve the application of leach solutions. Their objectives were to determine if sufficient permeability could be obtained for leaching. Fourteen programs fall into the experimental category, with four of them involving the Bureau of Mines. Data tables are only included for those sites where leach solutions were applied.

Bluebird Mine

The Bluebird mine is owned by Ranchers Exploration and Development Corp. and is located in the Miami-Globe area of Arizona about 75 miles (120 km) east of Phoenix. Production from the open pit mine began in late 1964, and the oxide ore is heap leached. The host rock is the Pinal Schist. The grade of the deposit averages 0.5 pct (*28*) and the principal copper mineral is chrysocolla with minor amounts of azurite and malachite. In 1968, Ranchers built the first commercial SX-EW plant at the mine to process the pregnant heap leach solutions. Copper production in 1980 was 12.2 million lb (5.5 X 10⁶ kg).

In 1979, a small-scale in situ leaching experiment was conducted in a section of the mine below the water table and under too much overburden to be considered for open pit mining. Vertical holes were drilled into the formation 100 ft (31 m) deep in a five-spot pattern with the corner holes spaced 20 ft (6 m) apart. Steel casings, 2-in diam (51-mm), were cemented in the four corner (injection) holes. The casings were perforated every 5 ft (1.5 m) and then each well was hydrofraced.

Ranchers officials have not publicly released any other information about the test except to say that the project has been "shelved" because it is not economic with current copper prices.

Ranchers has considered leaching in place the remaining ore in the pit. The ore body still contains 65 million T (59 X 10⁶ t) grading 0.53 pct Cu (*28*). This approach is being considered because it is no longer practical to remove the overburden required to mine the ore as an open pit.

Cerrillos Deposit

The Cerrillos deposit is located near Cerrillos, NM, about 25 miles (40 km) south of Santa Fe, and is owned by Occidental Minerals Corp. The deposit contains approximately 10 million T (9 X 10⁶ t) averaging 0.3 pct copper (0.2 pct acid soluble copper) and runs from the surface to 250 ft (76 m) in depth. The host rock is a granitic porphyry and chrysocolla is the principal copper mineral.

Occidental wanted to develop an in situ mining system for small oxidized copper deposits above the water table. A 100-ft (31-m) cube of surface ore was prepared by detonating 104,000 lb (47,200 kg) of AN-FO. Nine-in-diam (229-mm) blastholes spaced 13 ft (4 m) apart in a square pattern were used. The cube contained 80,000 T (72,600 t) of ore and had a surface area of 10,000 ft² (929 m²). The prepared surface of the block is shown in figure 10. Seventy percent of the material was reduced to minus 4 in (102 mm) in size by the blast.

A shaft was sunk to a depth of 120 ft (37 m) and a shotcreted drift was driven under the blasted ore. Five sets of 3-in-diam (76-mm) holes were fan drilled up into the blasted ore at 20-ft (6-m) intervals along the drift. These solution recovery holes varied in length from 25 to 58 ft (8 to 18 m) and were cased with 2-in-ID (51-mm) plastic pipe with 0.25-in (6-mm) perforations at 6-in (152-mm) intervals. No leaching was ever done, however, owing to changing environmental regulations which kept increasing costs. Occidental shut down the site in 1977 and is now in the process of selling it.

The site was to have been leached at a flow rate of 0.045 gpm/ft² (1.83 Lpm/m²) for a total flow of 450 gpm (1,700 Lpm) with pregnant solutions processed in a precipitation plant. The operation appeared technically feasible and a full-scale commercial operation had been planned.

Consolidated Copper Co. Mine

Consolidated Copper Co.'s Brooks ore body was located near Ely in east-central Nevada. Success by the Ohio Copper Co. in leaching its block caved workings prompted Consolidated to do experimental leaching in a portion of its block caved workings in 1925 (*6*). The ore was in a shear zone in monzonite with a fairly well-defined footwall. Copper mineralization was basically sulfide with very little oxide present. The ore body was developed by drifts and raises and caved through finger chutes down to the 360-ft (126-m) level. The ore as mined assayed 1.16 pct Cu, while the caved workings after mining were thought to contain about 0.3 pct Cu. The pregnant leach solution grade was about 1.0 gpL Cu (*4*). Operating data are shown in table A-2.

Emerald Isle Mine

El Paso Mining and Milling Co. and the Bureau of Mines conducted a cooperative research program at the Emerald Isle open pit copper mine from 1973 to 1975. The objective of the program was to develop in situ mining methods for ore exposed in the pit bottom and also for ore under 200 ft (61 m) of overburden adjacent to the pit. The mine is located in the northwestern portion of Arizona near the town of Kingman. Activities at the site included blasting and leaching 15,000 T (13,600 t) of ore in the pit bottom, followed by the leaching of 100,000 T (90,700 t) of unblasted pit-bottom ore. A

Figure 10.—Prepared surface of blasted cube at Cerrillos deposit.

fragmentation experiment under 200 ft (61 m) of overburden was also done. Two blasts were detonated and the fragmentation was analyzed by core drilling and permeability testing (*8*).

The host rock at the site is the Gila Conglomerate with an average thickness of 70 ft (21 m) and it dips 10° to 15° to the southwest. The principal copper mineral is chrysocolla with minor amounts of dioptase, tenorite, and cuprite. During the life of the open pit, 1.4 million T (1.27 x 10^6 t) of ore averaging 1.0 pct Cu was mined and the pit reached a depth of 200 ft (61 m). A cross section of the ore deposit showing the two test sites is shown in figure 11. The water table in the pit was maintained 5 ft (1.5 m) below the pit floor. In the overburden test area, the water table was 235 ft (72 m) below the surface (*8*).

A blast was detonated in the pit bottom to rubblize 15,000 T (13,600 t) of ore for leaching. Seven 8.375-in-diam (213-mm), 50-ft (15-m) blastholes spaced 25 ft (8 m) apart in a seven-spot pattern were used. The holes had a powder column averaging 22 ft (6.7 m), and 25 ft (7.6 m) of stemming. A total of 4,500 lb (2,043 kg) of slurry was detonated without delays. The powder factor was 0.3 lb/T (0.15 kg/t). Topographic surveys before and after the blast revealed a maximum surface rise of 1.4 ft (0.43 m). The blast had a significant effect on fragmentation with core recovery, rock quality designation (RQD), average piece length, and longest piece all decreasing from preblast to postblast core. The RQD is obtained by measuring the total length of all pieces of core greater than or equal to 4 in (102 mm) and dividing the total by the distance drilled (*36*).

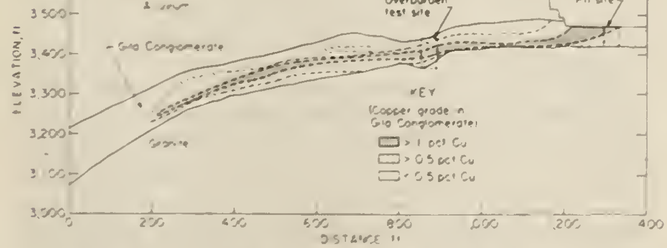

Figure 11.—Cross section through Emerald Isle mine.

Leaching of the blasted ore began in March 1974 and ran for 114 days. Dilute H_2SO_4 leach solution was distributed on the surface through perforated pipes and recovered through a well on the east side of the blasted zone. Pregnant leach solutions were processed in a precipitation plant. Total copper production during the test was 29,000 lb (13,150 kg). Operating data are shown in table A-2.

In December 1974, pit bottom leaching of about 100,000 T (90,700 t) of unblasted ore began and continued for 190 days. It was hoped that natural permeability would be sufficient for leaching. Seven recovery wells 50 ft (15 m) apart were used to collect the pregnant solutions. A total of 142,000 lb (64,500 kg) of copper was produced during the test (*8*).

Pit bottom leaching was stopped because the flow rates of leach solutions were not as high as desired, only 17 gpm (64 Lpm) per well. The highest production was from wells near the blasted portion of the pit. Flow rates per well ranged from 5 to 45 gpm (19 to 170 Lpm). A drilling and blasting program to improve permeability was started but not finished because all operations at the mine were closed down. Operating data are shown in table A-2.

Two test blasts were made through 200 ft (61 m) of overburden at a site outside the open pit (fig. 11). The first blast had seven 9-in-diam (229-mm) holes averaging 272 ft (85 m) deep. A seven-spot pattern was used with 20-ft (6-m) hole

spacings. A second blast of three holes spaced 18 ft (5.5 m) apart was necessary because the first did not create enough permeability for adequate leaching. The explosive used in both blasts was a smokeless powder slurry. Powder factors were 0.95 lb/T (0.48 kg/t) for the first blast and 1.47 lb/T (0.74 kg/t) for the second (*8*).

Both blasts had a significant effect on fragmentation, but permeability measurements were too low for adequate leaching. The two values measured after the second blast were 6 and 20 md. It is unknown whether these values were low because of insufficient blast-induced fracturing, or resealing of fractures. Another factor may have been the bentonite drill mud (used to drill the blast and core holes) which forms a low permeability wallcake on the borehole.

The Emerald Isle Mine was recently purchased by TRC Enterprises, Inc. Initial plans are to leach the pit bottom ore in situ when copper economics improve.

Johnson Mine

The Bureau of Mines and Cyprus Mines Corp. conducted a cooperative research program to investigate the in situ mining potential along the fringes of the open pit Johnson mine 17 miles (27 km) west of Willcox, AZ. This program led to a test blast detonated in August 1977, for evaluating blast design, solution flow rate, and solution containment (*7*). No in situ mining was done at the site.

The Johnson mine is an oxide copper deposit mined by open pit-heap leach methods. Pregnant leach solutions are processed in a SX-EW plant. The metashale ore is in the lower member of the Abrigo Formation and is underlain by 150 ft (46 m) of impervious Bolsa Quartzite. The principal copper mineral is chrysocolla with minor amounts of azurite and malachite. The grade of the deposit averages about 0.5 pct acid soluble copper. The ore has very low tensile strength [24 psi (172 kPa)], a porosity of 11 pct, and permeability of 1.7 md as measured on core samples. Field permeability ranges from 5 to 50 md (*7*).

A cross section through the mine is shown in figure 12. The pit at the time of the test and the proposed final pit limits are shown. The test blast area was 56 ft (17 m) below the surface and extended 185 to 224 ft (56 to 68 m) deep. Figure 13 shows the test blast design. Thirteen 9.875-in-diam (250-mm) blastholes were spaced 14 ft (4.3 m) apart in an equilateral triangle pattern. The pattern was elongated in the down-dip direction with the deepest holes on the northeast end. A total of 51,500 lb (23,380 kg) of AN-FO was loaded into the 13 blastholes. A constant stemming height of 56 ft (17 m) was maintained. The blast was bottom primed with two 0.4-lb (0.18-kg) cast primers. A powder factor of 2.2 lb/T (1.1 kg/t) fractured about 19,700 T (17,870 t) of ore assuming a 4-ft (1.2-m) overbreak. These figures are based on ore in the powder column only and not ore in the stemming

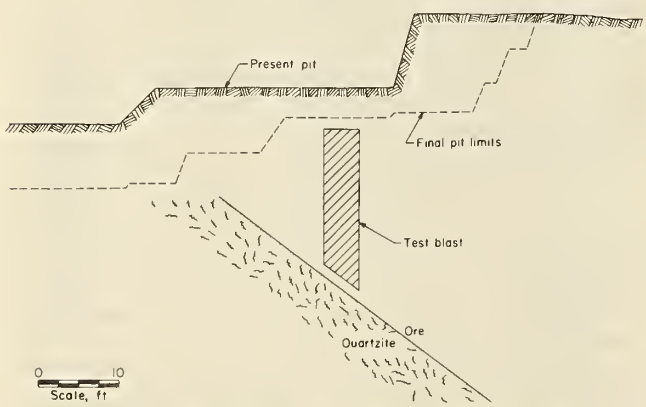

Figure 12.—Cross section of Johnson mine deposit.

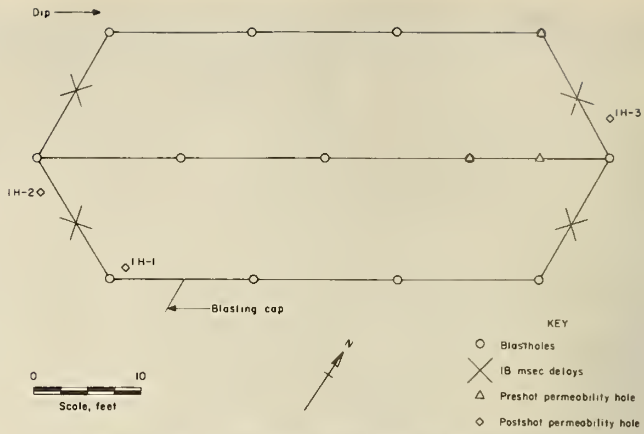

Figure 13.—Johnson mine test blast design.

region. The powder factor was higher than normally used in bench blasting but was considered necessary to break ore in a confined situation.

Topographic surveys were run before and after blasting. The surface rose over a broad area with significant displacement along a north-south-trending fault. The volume increase at the surface was 1,500 yd³ (1,150m³). The postshot drill core was very highly fractured with only 23 pct recovery and an RQD of 2 pct compared with 87 pct recovery and 51 pct RQD for preshot core (*7*).

Constant-head permeability tests were run before and after blasting. The test holes are shown in figure 13. Preshot permeabilities ranged from 15 to 43 md, while postshot values ranged from 180 to 8,500 md. These postshot values are considered adequate for successful leaching.

A water circulation test was run by injecting water into holes IH-1 and IH-2 and by monitoring the water in IH-3 (fig. 13). During this test 300,000 gal (1.4×10^6 L) of water were injected at 35 gpm (132 Lpm) in an attempt to fill the bottom of the fractured zone. However, very little water was observed in the 224-ft-deep (68-m) IH-3 hole. The water may have escaped from the fractured zone along the Bolsa Quartzite contact, or the IH-3 hole may not have been deep enough and insufficient water was injected to fill all the void spaces created at the bottom of the fracture zone. Based on this test, it was determined that a solution containment system, such as grouting, would be necessary for successful leaching.

Kimbley Pit

The Kimbley pit is located in east-central Nevada, near the town of Ruth, and is part of Kennecott Copper Corp.'s Nevada Mines Division. It is one of several small open pits in the area and has been worked to its economic limit. The host rock has been tentatively identified as a limey sediment which has been intruded by biotite-argillic porphyry (*35*). The principal copper mineral is chalcocite and the ore contains significant amounts of pyrite.

Two preliminary permeability tests and a pilot scale in situ leach mining test were done at the site. The objectives of the preliminary tests were the determination of formation permeability and extraction of copper solutions from the formation (*35*).

The first test was conducted in 1968 on a bench 250 ft (76 m) above the pit bottom. Five injection holes were drilled. They were 9-in diam (229-mm), 40 ft (12 m) deep, 50 ft (15 m) apart, and 25 ft (8 m) from the edge of the bench. Acidified water was pumped into the holes for 2 weeks. The holes were kept full of solution, but not injected under pressure. The average permeability at the start of the test was 38.4 md, but by the end of the test it had risen to 268.8 md. Subsequent tunneling and blasting revealed that solutions radiated from

the holes in a cone shape that reached 40 ft (12 m) in diameter at a depth of 165 ft (50 m) below the top of the hole (*35*).

The second test was done at a site 1,000 ft (305 m) east of the pit. A series of 100-ft (31-m) vertical injection holes were drilled from the surface above an old exploration drift. Leach solution injection into the holes produced a cone of influence 160 ft (49 m) in diameter. Permeability averaged 6,209 md. The drift intercepted about 7 pct of the injected solution (*35*).

The pilot leaching test was conducted in the exploration drift used for a second test. A recovery well was drilled from the surface 265 ft (81 m) deep into a bowl-like limestone formation underlying the ore. The drift was modified to facilitate leach solution application. Dams were built to contain the solution in a portion of the drift, and holes were drilled for solution distribution to supplement the natural fracture system. Beginning in January 1970, acidified fresh water was pumped between the dams in the drift at the rate of 50 gpm (190 Lpm). The solution was allowed to percolate down through the natural fractures in the ore, and migrate to the recovery well in the limestone basin to be pumped to the surface.

Injection was continued for 17 months. Only about 0.2 gpm (0.8 Lpm) of solution was migrating to the well with an average grade of 0.15 gpL Cu. Both solution flow and grade were uneconomic and the test was discontinued. Experimentation at the site continued through 1972, however, with efforts involving hydraulic fracturing, liquid oxygen injection, and acidified ferric sulfate leach solution. The results of these tests have not been published. Operating data are shown in table A-2.

Lakeshore Mine

The Lakeshore mine is located 30 mi (48 km) southwest of Casa Grande, AZ. The property contains three copper bearing bodies; two sulfide and one oxide. Hecla Mining Co. initially developed the property (*25*) in the late 1960's as a block caving operation in both sulfide and oxide ore. The mine was sold to Noranda Lakeshore Mines, Inc., in 1979 after the copper recession of the mid-1970's forced Hecla to abandon the project (*26*). Noranda continued with conventional underground mining and vat leaching of the oxide ore, and opened a new SX-EW plant in 1981 to process the pregnant vat leach solutions.

Low copper prices are again causing a planned shutdown of underground mining for late 1983. At that time a complete changeover to in situ mining is scheduled to have taken place. This will be cheaper, require fewer people, and take advantage of equipment already at the site (*26*).

Experimental work for the in situ mining began in early 1983. Leaching is being done in block caved areas. Dilute sulfuric acid leach solution is applied to the ore through injection holes from the surface. They have been drilled in the subsidence area that resulted from the caving operation, average 550 ft (168 m) in depth, and are cased. Five dams built in the underground workings are used to collect the pregnant leach solution. From there it is pumped to the SX-EW plant on the surface. Detailed operating data are not yet available.

Medler Mine

Perhaps the first attempt at in situ copper mining took place between 1906 and about 1909 at the Medler mine near Clifton, AZ (*2*). The ore was the primary sulfide portion of a porphyry copper deposit. It had an average grade of 0.38 pct Cu which was considered at the time to be too low for conventional mining.

At the time the mine was leased for this experimental work, it consisted of two adits vertically separated by 60 ft (18 m) with a connecting winze. Development work for leaching consisted of driving another adit 100 ft (30 m) above the upper level and sinking the winze 100 ft (30 m) to get below the water table. At the bottom of the winze, another level was projected to facilitate collection of the drainage water (*2*). Drifts were driven off the main adits on all levels at intervals no greater than 200 ft (61 m) for water application and recovery.

Leaching was done by flooding the second level with mine water and allowing it to seep down to the third level where it was collected and passed through cementation launders. The launders were located in the third level adit. Makeup water was bailed from the fourth level.

Initial water seepage was very slow. It took months for any water to show up on the third level which was only 60 ft (18 m) below the second. Once the rock was saturated, however, the flow rate increased. Pregnant leach solution grades ranged from 0.2 to 0.6 gpL. The first carload of copper precipitate was shipped in July 1908, 2 yr after development work was begun (*2*). The operation was closed by the right of eminent domain when a railroad tunnel was driven through the mine.

Mountain City Mine

The Mountain City mine is located about 7 miles (11 km) southwest of Mountain City, NV. It was first operated by the Anaconda Copper Co., as the Rio Tinto mine, from 1932 to 1948 with copper ore averaging 9.75 pct being mined by square set stoping methods. The most recent in situ copper leaching activities took place from 1972 to 1974 when the property was co-owned by Cliffs Copper Corp., a subsidiary of Cleveland-Cliffs Iron Co. and E.I. du Pont de Nemours & Co. (*4*).

The ore occurs in a stratigraphically restricted zone of black and gray phyllite with associated quartzite lenses in a shale sequence called the Valmy formation. The major copper mineralization is supergene chalcocite which was deposited near the water table, 200 ft (61 m) below the surface. Minor copper minerals are chalcopyrite, cuprite, malachite, chrysocolla, azurite, and native copper. The mineralization was entirely along fracture planes and as replacement or coatings on disseminated pyrite.

Copper was recovered from the Rio Tinto mine from August 1966 through February 1970 through a leaching process in which solution was withdrawn from the flooded mine, passed over scrap iron to precipitate the copper values, acidified with sulfuric acid to about pH 2, and returned to the mine (*6*).

Initial operations distributed barren leach liquor to four holes that entered the mine at the 222-ft (68-m) level. Some leach liquor was also added to four subsidence pits above the ore body. Three additional holes entering the 300-ft (92-m) level were drilled and activated in early 1969. Approximately one-third of the barren liquor was returned directly to the mine shaft because of the limited capacity of the water injection holes.

Compressed air [80 to 100 psi (551 to 689 kPa)] was fed through two holes entering the 400-ft (122-m) level of the mine (about 150 to 200 ft (46 to 61 m) below the water surface) during 1968 to 1969 to assist oxidation and the leaching process (*6*).

During summer and fall months, copper was also leached from a nearby tailings pond. Pregnant liquor from the tailings operation was combined with pregnant liquor from the mine ahead of the cementation plant. During these periods, the amount of copper in the plant effluent solution increased as the loading on the cementation plant increased. Pregnant liquor grade from the mine usually increased an equivalent amount because the effluent was recirculated (*6*).

Operations were suspended in February 1970 when the supply of tailings had been exhausted and the grade of pregnant liquor from the mine was too low to support further operations.

Preparations for in situ leaching began again in 1972 when dewatering of the old workings began. This was fol-

lowed by shaft rehabilitation. The plan was to develop an unmined portion of footwall ore, known as the South Ore Body, by block caving, and then leach the caved areas. This method was determined to be the only economic way to recovering copper from the site.

Two test blocks were developed and caved. The first, 140 by 150 by 200 ft high (43 by 46 by 61 m) contained 306,000 T (277,600 t) and averaged 1.1 pct Cu. The second block contained 487,000 T (441,800 t) and averaged 0.93 pct Cu. Development work was done at the 400-ft (122-m) level. The caving procedure was based on normal long-hole drilling and block-caving methods. A draw of 12.5 pct was tried on the first block, but this caused undesirable caving above the 200-ft (61-m) level. The draw on the second block was decreased to 6 pct, which was estimated to be the minimum amount necessary to induce the desired degree of caving. Draw material was placed in heaps on the surface and leached (*4*).

The leach solution was a combination of H_2SO_4 and recycled mine water, which had been routed through a tailings area to become enriched in ferric sulfate. Solutions were introduced to the caved blocks through injection wells from the surface. They were initially drilled on 50-ft (15-m) centers; however, collapsed casings and low injection rates required drilling on 25-ft (8-m) centers. The injection rate was initially 25 gpm (95 Lpm) per hole which declined in 1 to 2 months to 0 to 5 gpm (0 to 19 Lpm) because of casing collapse and silt buildup. This resulted in gross variations in the total flow (*4*).

Solutions percolated down through the caved ore to the 400-ft (122-m) level, and collected with mine seepage at the 450-ft (137-m) level. Solutions were pumped to the precipitation plant with a submersible pump.

Leach solution was first applied to the first block in March of 1974 and the second block in May 1974. After solving initial startup problems, the blocks were continuously leached from June through November 1974. The pregnant leach solution at the 400-ft (122-m) level graded 1.5 to 2.0 gpL Cu, but after dilution with mine seepage the grade to the precipitation plant was 0.5 to 0.6 gpL. During the 6 months of continuous operation, a total of 850,000 lb (385,900 kg) of copper was produced from the two blocks, about a 5.5-pct recovery (*4*).

In December 1974 all activities at the mine were terminated because of higher than expected development costs and low copper prices [$0.55/lb ($1.21/kg)]. There were also unresolved technical problems with solution distribution and aeration which resulted in a slower than anticipated leaching rate for the copper sulfide minerals. The mine was sold to another company, which in turn, sold it again. Operating data are shown in the table A-2.

Nacimiento Mine

The Nacimiento mine is located about 4 miles (6.4 km) southeast of Cuba, in north-central New Mexico. In 1971, Earth Resources Co. began an open pit mining operation at the site which continued 3.5 yr. The deposit had estimated reserves of 9.6 million T (8.7 X 10^6 t) of ore averaging 0.66 pct Cu. Mine production totaled 2.5 million T (2.3 x 10^6 t) with a cutoff grade of 0.3 pct. The mine closed because of low copper prices and slope stability problems.

The host rock is the Agua Zarca Sandstone, which is 120 ft (37 m) thick and is overlain by shale and underlain by mudstone. Both of these confining layers are impervious. Permeability of the sandstone ranges from 300 to 3,000 md, which is sufficient for in situ leaching without blasting. The formations dip 29° to 34° to the southwest, and the principal copper mineral is chalcocite. All of the ore considered for leaching is below the water table which is at the bottom of the open pit. Column leaching tests on ore samples showed that ferric sulfate leach solution yielded excellent recovery,

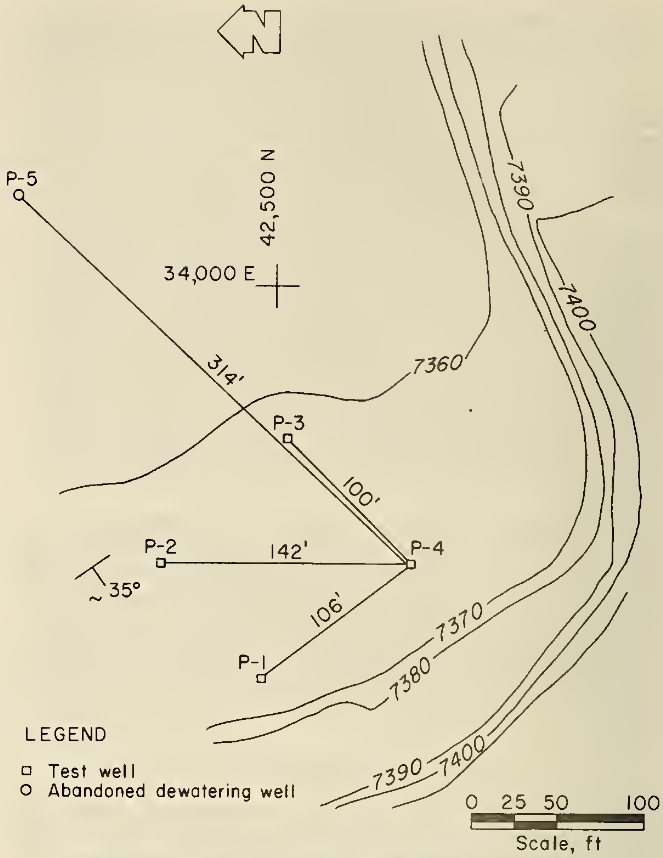

Figure 14.—Nacimiento mine well location map.

although oxygen may have to be added for leaching below the water table.

Hydrology tests were run in 1978. The test site was located on a bench in the southwest side of the pit. Four wells were drilled in a square pattern as shown in figure 14. The wells were 100 to 110 ft (30 to 40 m) deep and were cased with 4-in-diam (102-mm) plastic pipe; the sandstone ore layer was screened through at least 80 pct of its thickness. An abandoned dewatering well was also used. Drawdown and recovery tests were run, with pumping done from well P-4 and water levels monitored at all wells (*39*). The test results were "positive," proving a favorable hydrology for in situ leaching, but no further development has taken place because of low copper prices. Earth Resources still controls and maintains the property and plans to continue with the program when copper economics improve.

Safford Deposit

The Safford deposit is located 9 miles (14 km) north of the town of Safford, AZ, which is about 80 miles (130 km) northeast of Tucson. The host rocks for the deposit are andesitic volcanics. Principal copper minerals are chrysocolla, chalcopyrite, brochantite, chalcocite, and covellite with minor amounts of bornite. The deposit contains approximately 2 billion T (1.8 X 10^9 t) averaging 0.41 pct Cu, about half of which is relatively enriched oxide ore. The deposit is up to 4,000 ft (1,220 m) long and about 1,600 ft (488 m) thick and is overlain by 500 to 1,300 ft (150 to 400 m) of leached capping and barren volcanics (*10*). It is above any known water table.

Kennecott Minerals Co. purchased the property in 1959 after 4 yr of exploration. In the mid-1960's it proposed Project Sloop, which was to have been a joint effort with the

Atomic Energy Commission to fragment a portion of the ore body with a nuclear explosive for in situ mining (*10*). In the early 1970's Project Sloop was canceled because of perceived environmental restrictions.

Research was then redirected to conventional means for fragmenting the ore. The goal was to develop a system for in situ leaching the sulfide ore in the deposit. The research effort covered many areas including mineralogy, petrology, leaching chemistry, laboratory leaching, permeability testing, geophysical logging, ground water tracers, directional drilling, blasting, five-spot pattern leaching, plant design and economic analysis. It was one of the most intensive and expensive research programs to date, and the results are proprietary. However, several patents concerning various aspects of deep in situ mining were granted (*9, 14-16*).

Currently there is no research being conducted at Safford because of disappointing test results and the low price of copper. However, Kennecott plans to continue research in the future.

Seneca Mine

The Bureau, in cooperation with Homestake Copper Co., performed a confined blasting test in an underground native copper mine in the Keweenaw Peninsula of upper Michigan. It was done to determine the feasibility of using such blasting (with no relief for expansion) to fragment deep ore bodies for leaching (*5*). The test was done in 1976 in the abandoned underground workings of the Seneca mine near Mohawk, MI.

The geology of the district is a series of basaltic flows, often interspersed with conglomerate beds, which were tilted subsequent to their extrusion and now dip toward Lake Superior. In the mine, the copper occurred in the amygdaloidal top of the Kearsarge Basalt, which dips about 37° at the mine.

The basic approach to confined blasting was to drill blastholes downdip into the formation from the third level drift, 280 ft (85 m) below the surface. Parameters such as hole depth, hole diameter, hole spacing, explosive type, and stemming were considered in designing a blast that would break rock without moving it. Hole depth was set at 40 ft (12 m) to minimize the effects of the drift on the blast yet keep drilling costs within the research budget. A 3-in-diam (76-mm) hole was then selected based on the expected range of burdens associated with the 10-ft (3-m) ore zone thickness. Finally, a slurry blasting agent suitable for wet conditions was chosen for the shots with a 20-ft (6-m) powder column and 20 ft (6 m) of water stemming (*5*).

The only significant blasting parameter remaining for evaluation in the tests was blasthole spacing, which held the key to economic feasibility. If the blastholes had to be drilled very close together to create adequate permeability, confined blasting would be uneconomical. To evaluate fracturing and permeability as a function of blasthole spacing, a blast incorporating three different triangular patterns was designed (fig. 15).

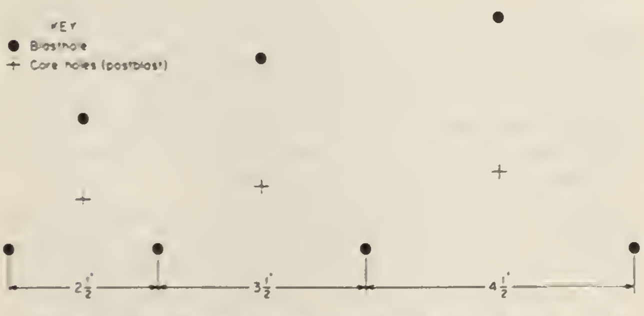

Figure 15.—Seneca mine blasthole pattern in drift wall.

The three burden to blasthole diameter ratios selected for the test series were 10, 14, and 18. These ratios corresponded to blasthole spacings of 2.5, 3.5, and 4.5 ft (0.76, 1.07, and 1.39 m). The calculated powder factors in the ore around the loaded portions of the blastholes were 8, 4, and 2.5 lb/T (4, 2, and 1.2 kg/t) respectively (*5*). Core obtained from these holes was analyzed for fractures; permeability tests were also conducted in the blastholes before shooting.

The fragmentation effectiveness was evaluated with postshot core holes drilled in the center of each triangular pattern. Fracturing in the core was then compared with that from the preshot core; formation permeability was also compared.

All preshot tests and analyses pointed to a "tight" impermeable formation that contained few fractures. Crude measurements of permeability ranged from 0 to 1.5 md. Blasting did create new fractures, particularly at the closer spacings with their associated high powder factors, but the permeability changes created by the fracturing were small. Even with injection pressures of 80 psi (552 kPa) very little fluid was transmitted from one hole to the next. Because the best permeability achieved with the close spacings was only 40 md, confined blasting simply did not create enough permeability to consider leaching.

Sierrita Mine

The Bureau of Mines performed a fragmentation experiment at the Duval Corp. Sierrita open pit copper-molybdenum mine in 1973. The mine is about 24 miles (39 km) south of Tucson, AZ, in Pima County. The experiment consisted of a preblast evaluation of the site, drilling and blasting, and postblast evaluation. No leaching was done at the site (*36*).

The dominant rock type was well-weathered quartz monzonite porphyry that had considerable jointing and faulting. Assays of samples taken during blasthole drilling averaged 0.14 pct Cu, with 0.13 pct acid soluble.

Preblast evaluation was done by diamond core drilling three NX size [approximately 3-in-diam (76-mm)] holes in the middle of the 15-, 20-, and 25-ft (4.6-, 6.1-, and 7.6-m) blasthole spacing areas as shown in figure 16. The 10 blasthole locations and blast delay sequence are also shown in this figure. The 9-in-diam (229-mm) blastholes were 110 ft (33.5 m) deep. Each hole had a powder column of 50 ft (15 m) and 60 ft (18 m) of stemming. The blast contained a total of 17,440 lb (7,918 kg) of 10 pct aluminized slurry blasting agent. Powder factors, assuming infinite patterns, were 0.40, 0.63, and 1.13 lb/T (0.20, 0.32, and 0.57 kg/t) for the 25-, 20-, and 15-ft (7.6-, 6.1-, and 4.6-m) patterns, respectively (*36*).

Postblast studies included a topographic survey, mapping of surface fractures, fragment size distribution

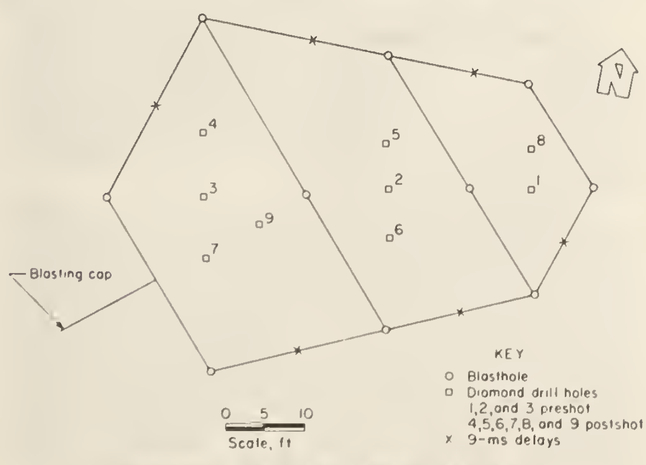

Figure 16.—Sierrita mine test blast design.

Table 2.—Sierrita mine drill core data

Location	Hole	Length recovery, pct	Weight recovery, lb/ft	RQD, pct	Largest piece, in	Average size of pieces >1 in, in
Preshot	1	101	2.63	37	13	3.1
Do	2	98	2.58	35	17	3.2
Do	3	96	2.58	37	12	3.2
25-ft pattern	4	81	2.08	28	10	2.8
Do	7	76	1.90	19	11	2.8
25-ft pattern (off center)	9	62	1.61	19	12	2.7
20-ft pattern	5	53	1.20	9	8	2.2
Do	6	51	1.20	12	9	2.5
15-ft pattern	8	37	.99	10	9	2.3

measurements of surface material, and six NX core holes located as shown in figure 16. Laboratory measurements done on preblast and postblast cores were recovery by length and weight, RQD, and fragment size distribution.

The surface rise over the blast averaged about 5 ft (1.5 m), and the total volume increase produced by the blast was 5,100 yd³ (3,900 m³). Table 2 lists the location, length core recovery, weight core recovery, RQD, largest piece, and average size of pieces greater than 1 in (25 mm) for nine diamond-drill core holes. Length core recovery and weight core recovery were measured after the core was returned to the laboratory. The length core recoveries measured in the core boxes were higher (101 pct for hole 1) than would have been obtained from measurements taken before the core was removed from the core barrel. Although there is a good correlation between length and weight recovery, the weight core recovery was considered the most accurate method for evaluating blast damage. The average size of core pieces 1 in (25 mm) or greater was determined by dividing the total length of all pieces greater than 1 in (25 mm) by the number of pieces greater than 1 in (25 mm).

The drill core data show that better breakage occurs as blasthole spacing decreases, and fragmentation improves as distance from the center of the equilateral triangle patterns increases (see hole 9 in table 2). Factors other than fragmentation affect copper recovery, and actual leaching would have been desirable. However, the fragmentation analysis indicates that all three blast patterns produced adequate breakage for in situ mining.

Van Dyke Deposit

The Van Dyke deposit partially underlies the town of Miami, AZ, which is about 75 miles (120 km) east of Phoenix. The deposit contains approximately 100 million T (90 X 10⁶ t) of ore with an average grade of 0.5 pct Cu. It dips about 15° and ranges from 1,100 to 2,000 ft (335 to 610 m) deep beneath the town and is overlain by the Gila Conglomerate (*27*). Copper mineralization occurs in hairline fractures in the Pinal Schist (*22*). The major copper mineral is chrysocolla with minor amounts of azurite and malachite.

Occidental Minerals Corp. purchased the lease for the property in 1968 and spent $11 million on exploration and tests between then and 1980 (*27*), when it developed a plan for mining the ore body in situ. Conventional mining methods were not considered viable because of deposit depth, location, and grade.

Two in situ mining tests were conducted below the water table with injection and recovery wells drilled from the surface. The first test, in 1976, involved two wells 75 ft (23 m) apart and just over 1,000 ft (305 m) deep (*22*). The 10-in-diam (254-mm) holes were drilled and cased with 4-in-diam (102-mm) fiberglass casing. Both wells were then hydrofraced. This fracture was supposed to extend a maximum of 200 ft (61 m) from the wells. After the casings were pressure tested for leaks, dilute H_2SO_4 leach solution was injected into one well, and production of solution from the other well began. Injection and production were done at the same rate. Pregnant leach solutions produced during the test were transported to a nearby SX-EW plant for processing.

A second leach test was then conducted using a five-spot pattern with 100-ft (31-m) well spacings. Five 10-in-diam (254-mm) holes were drilled and cased with 4-in (102-mm) fiberglass casing. The test zone was between 1,000 and 1,200 ft (305 and 366 m) deep. All of the wells were hydrofraced. Dilute H_2SO_4 solution was injected in the center well and pregnant solutions were produced from the corners. The test ran for several months and was considered successful. Details of the testing are proprietary.

At this point, Occidental applied for two special-use permits from the town of Miami to do further testing in another portion of the ore body, but the permits were denied. After a long and costly legal battle over the permits, Occidental dropped its lease on the property in October 1980 (*24*). There is currently no activity at the site.

The full-scale commercial operation planned by Occidental would have involved a shaft outside of town, and a gridwork of nearly horizontal drifts above the ore body. Injection and production wells would have been drilled from the drifts. A SX-EW plant was also planned. The operation would have employed about 200 people and had an estimated life of 15 yr (*22*). Operating data for the leaching tests are shown in table A-2.

SUMMARY

To date, there have been at least 24 sites in the United States where in situ copper mining production or research has taken place. Three have been commercial operations with ore body preparation (blasting), seven have been commercial operations in old mine workings, and 14 were experimental sites. Of these experimental sites, only eight reached the leach-solution application stage.

Ore bodies that are too low grade or small to be mined by conventional methods have potential for in situ mining because of the relatively low capital and production costs involved. The method is not normally considered for other deposits because of its relatively low recovery. Leaching of old mine workings such as block caves, backfilled stopes, and open pits, is often an economic method for recovering additional copper when conventionally minable reserves have been depleted. Experimental work has typically been done on a small scale to determine the feasibility of commercial operations, however, several of the Bureau of Mines experiments were directed specifically at ore fragmentation by blasting and were not leached.

At present the only two active, commercial, in situ copper operations in the United States (Copper Queen Branch and Miami mine) involve leaching of old mine workings. The Lakeshore mine is the only active experimental site, and Noranda is planning to go commercial in the near future.

In situ copper mining activities have been shut down for several reasons, with the most common one being economic (low copper prices, excessive ore preparation costs, etc.). Other causes of shutdowns have been expansion of open pit operations into leach areas, environmental concerns, technical difficulties (inability to contain solutions, low pregnant solution grade, etc.), and one operation was closed when the lease agreement was not renewed.

Many of the companies plan to resume leaching programs when copper economics improve, and Ranchers is considering a plan to blast and leach in situ, the remaining ore in the Bluebird open pit. In situ mining will become much more attractive and play an increasingly important role in future copper production as conventional mining costs increase and ore grades decrease.

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APPENDIX A.—IN SITU COPPER MINING OPERATING DATA

Table A-1.—Commercial operations

Operation	Big Mika Mina	Old Raliabla Mina	Zonia mina	Burro Mountain Branch
Location	Winnamucca, NV.	Copper Craak naar Mammoth, AZ.	Kirkland, AZ.	Tyrona, NM.
Host rock	Greenstona, chart, argillita, soma limastona.	Volcanic agglomarata, primarily andasita.	Shala, andasita.	Quartz monzonita, porphyry, granita.
Copper minarals:				
Principal	Cuprita, tanorite, chalcopyrita.	Chalcocita, chalcopyrita, malachita, chrysocolla.	Chrysocolla.	Chalcocita.
Minor	Not availabla.	Chalcanthita, azurita, nativa.	Azurita, malachita, tanorita.	Chrysocolla, covallita, chalcopyrita, bornita, azurita, malachita.
Avaraga ora grada, pct	1.18	0.84	0.2 (blastad portion of daposit only).	Not availabla.
Ora praparation	Blasted pit walls and bottom with angled and vertical blastholas, tarracad.	Coyota blast of hillsida using old and naw underground mina workings, terracad.	Blasted pit walls and bottom with vartical blastholas (3 saparate blasts).	Nona, laachad block-cavad araas.
Araa involved, ft²	122,000	245,000	390,000 (1st blast only).	Not available.
Maximum dapth, ft	160	500	240	Do.
Quantity of matarial, T	640,000	4 million.	6,830,000	Do.
Avarage fragmant size, in	9	8.5	12-14	Do.
Laach solution	Diluta H_2SO_4.	Diluta H_2SO_4.	Diluta H_2SO_4.	Watar.
pH	1.6-2.0	1.7-2.3	2.0-2.2	Not availabla.
Solution application	Rainbird sprinklars.	Rainbird sprinklers 1st laach, sprinklars and 6 injection holas 50 ft deep 2d laach.	Rainbird sprinklars.	Do.
Flow rata, gpm	175-200	1,000-1,100 (1st laach), 600-700 (2d laach).	500-1,000	Do.
Solution recovery	Recovary wall (16-in diam, 180 ft deep).	Flowed from hillsida at basa of rubbla and collected in basin.	Racovary wall in pit bottom (10-in diam, 100 ft daap).	Undarground workings.
Watar table	110 ft below postblast surfaca.	1-8 ft below lowast adit.	Prablast pit bottom.	Not availabla.
Pregnant solution copper contant, gpL	Initial 2.0, than dropped to 1.0 1st laach, 0.8 2d leach.	Initially 1.8, than dropped to 0.8 at 1st shutdown, 0.4 at 2d shutdown.	2.0-0.8	Do.
Type of copper recovary	Pracipitation on scrap iron.	Pracipitation on scrap iron.	Pracipitation on scrap iron.	Pracipitation on scrap iron.
Consumption, lb/lb Cu:				
Acid (H_2SO_4)	1.5	5.0	15	Not applicabla.
Iron	1.75	2.4	2.1	Not availabla.
Copper production, lb/d	5,000 (maximum).	20,000 (dasign capacity).	5,000 (maximum).	Do.
Employees	6-8	15	25	Do.
Activa datas	1973-74, 1978-79	July 1972-Saptambar 1974, August 1979-January 1981	April 1973-early 1975	May 1941-1949

Operation	Butta Montana mines	Coppar Quaan Branch	Inspiration mina	Miami mina
Location	Butta, MT.	Bisbee, AZ.	Claypool, AZ.	Miami, AZ.
Host rock	Quartz monzonite.	Sacramanto quartz porphyry.	Granita schist.	Pinal Schist.
Copper minarals:				
Principal	Chalcopyrita, chalcocita.	Chalcocite.	Azurite, malachita, chrysocolla.	Chalcocita.
Minor	Bornita, azurite, malachita.	Azurita, malachita.	Chalcopyrita, chalcocita, cuprita, covellita.	Covallita, malachita, azurita, bornita, chalcopyrita, chrysocolla, cuprita, nativa.
Avarage ora grada, pct	0.8	0.29	0.5	0.88 original ora body, cavad stopas unknown.
Ora praparation	Nona, laached backfillad stopes.	Nona, laached open pit and undarground workings.	Nona, laachad block-caved stopes.	Glory hola ovar block-cavad araa tarracad.
Araa involved, ft²	Not availabla.	Not availabla.	130,000-175,000	5 million.
Maximum dapth, ft	5,000	1,800	850	1,000 (600 ft of cavad matarial).
Quantity of matarial, T	Not availabla.	Not availabla.	Not availabla.	Not availabla.
Avarage fragmant siza, in	Do.	Do.	Do.	Do.
Laach solution	Mina watar with a small amount of H_2SO_4.	Mina watar.	Diluta H_2SO_4.	Diluta H_2SO_4.
pH	2.1	7	Not availabla.	1.5
Solution application	Old access to stopes or injection holas (3-in diam, 10 ft apart).	Rotating haad sprinklars.	Injaction holas (12-in diam, 250-300 ft deep).	Parforatad pipe sprays and injaction holas (6-in diam, avaraga dapth 195 ft).
Flow rata, gpm	700 (maximum).	500	190-560 (yaarly avaragas).	3,070
Solution recovary	Undarground workings.	Undarground workings on 1,800-ft laval.	Undarground workings on 850-ft laval.	Dams in drifts on 1,000-ft laval.
Watar tabla	Maintained at lowast working laval.	Maintainad at 3,100-ft laval.	Abova watar tabla.	Not availabla.
Pregnant solution copper contant, gpL	0.50-0.75	0.6 (mixad in situ and dump solutions).	1.8 initially, 0.7 last yaar of operation.	0.835 (1981).
Type of copper recovary	Pracipitation on scrap iron.	Precipitation on scrap iron.	Precipitation on scrap iron.	Precipitation on scrap iron until 1976, prasently SX-EW.
Consumption, lb/lb Cu:				
Acid (H_2SO_4)	Not availabla.	Not applicabla.	Not availabla.	3.2 (4.0 pre-1976).
Iron	Do.	Not availabla.	2.5	2.0 pre-1976.
Copper production, lb/d	33,000 (avaraga).	5,800 (maximum in situ production).	5,200 (avaraga).	30,000-35,000 dasign capacity.
Employees	15-30 (undarground craw only).	2 (pit plumbing needs only).	1-4 (solution application only).	30-35
Activa datas	1930's-1964	1975 to prasent.	1967-74	1942 to prasant.

Table A-1.—Commercial operations—Continued

Operation	Ohio Copper Co. mine	Ray mine
Location	Bingham Canyon, UT.	Ray, AZ.
Host rock	Quartzite with quartz monzonite intrusion.	Schist and diabase.
Copper minerals:		
Principal	Chalcocite.	Chalcocite.
Minor	Malachite, azurite.	Chrysocolla, malachite, azurite.
Average ore grade ... pct	0.3	1.0
Ore preparation	None, leached block-caved material.	None, leached block-caved material.
Area involved ... ft²	840,000	435,000
Maximum depth ... ft	1,900 (1,200 ft of caved material)	300
Quantity of material ... T	38 million.	Not available.
Average fragment size ... in	4	Do.
Leach solution	Creek and mine water.	Fresh water.
pH	Not available.	7
Solution application	150-ft launder with 2-in-diam holes, moved on a regular pattern.	Rotating sprinklers.
Flow rate ... gpm	1,200-1,400	340-500
Solution recovery	Tunnel on 1,900-ft level.	Concrete dams in underground drifts.
Water table	Above water table.	Above water table.
Pregnant solution copper content ... gpL	2.02	9.23
Type of copper recovery	Precipitation on scrap iron.	Precipitation on scrap iron.
Consumption, lb/lb Cu:		
Acid (H_2SO_4)	Not applicable.	Not applicable.
Iron	1.0	1.15
Copper production ... lb/d	20,000 (average).	20,000 (average).
Employees	Not available.	Not available.
Active dates	Started in 1922, now part of Bingham Canyon open pit.	Started in 1937, now part of Ray open pit.

Table A-2.—Experimental operations involving leach solution application

	Consolidated Copper Co. mine
Location	Ely, NV.
Host rock	Shear zone is monzonite.
Copper minerals:	
Principal	Sulfide.
Minor	Very little oxide.
Average ore grade ... pct	0.3
Ore preparation	None, leached block-caved material.
Area involved ... ft²	Not available.
Maximum depth ... ft	360
Quantity of material ... T	Not available.
Average fragment size ... in	Do.
Leach solution	Water.
pH	Not available.
Solution application	Do.
Flow rate ... gpl	Do.
Solution recovery	Underground workings
Water table	Not available.
Pregnant solution copper content ... gpL	1.0
Type of copper recovery	Precipitation on scrap iron.
Consumption, lb/lb Cu:	
Acid (H_2SO_4)	Not applicable.
Iron	Not available.
Copper production ... lb/d	Do.
Employees	Do.
Active dates	Started in 1925, closure date unknown

Table A-2.—Experimental operations involving leach solution application—Continued

Operation	Emerald Isle mine		Kimbley pit	Medler mine
	Pit blast leach	Pit bottom leach		
Location	Kingman, AZ.	Kingman, AZ.	Ruth, NV.	Clifton, AZ.
Host rock	Gila Conglomerate.	Gila Conglomerate.	Limey sediment intruded by argillic porphyry.	Porphyry.
Copper minerals:				
Principal	Chrysocolla.	Chrysocolla.	Chalcocite.	Sulfide.
Minor	Dioptase, tenorite, and cuprite.	Dioptase, tenorite, and cuprite.	Not available.	Oxide.
Average ore grade ... pct	1.0	1.0	0.32	0.38
Ore preparation	Blasted pit bottom with vertical blastholes.	None, leached pit bottom.	None.	None, drifts driven for access.
Area involved ... ft²	4,420	28,125	1,200	Not available.
Maximum depth ... ft	50	50	265 (165-ft ore thickness).	260
Quantity of material ... T	15,000	100,000	12,000	Not available.
Average fragment size ... in	Not available.	Not available.	Not available.	Not applicable.
Leach solution	Dilute H_2SO_4.	Dilute H_2SO_4.	Dilute H_2SO_4.	Water from mine sump.
pH	1.18	1.10	Not available.	Not available.
Solution application	Perforated pipes.	Perforated pipes.	Drift injection (through fractures and drill holes).	Flooded drifts.
Flow rate ... gpm	63.5	118	50 injected, 0.2 recovered.	Not available.
Solution recovery	Recovery well (11-in diam, 50 ft deep, cased with 10-in-diam PVC pipe perforated in bottom 20 ft.)	7 recovery wells (12-in diam, 50 ft apart, 180 ft deep).	Recovery well (265 ft deep, at low point in limestone basin under test area).	Drifts.
Water table	5 ft below pit bottom.	5 ft below pit bottom.	170 ft below surface.	Between 3d and 4th mine levels.
Pregnant solution copper content ... gpL	0.59	0.65	0.08-0.17 (0.15 average).	0.2-0.60
Type of copper recovery	Precipitation on scrap iron.	Precipitation on scrap iron.	Not available.	Precipitation on scrap iron.
Consumption, lb/lb Cu:				
Acid (H_2SO_4)	15.0	10.0	Do.	Not applicable.
Iron	4.7	2.75	Do.	Not available.
Copper production ... lb/d	245 (average).	748 (average)	Do.	Do.
Employees	Not available.	Not available.	Do.	3 (solution handling only).
Active dates	March 1974-June 1974.	December 1974-June 1975.	1970-71 (pilot leach test only).	April 1906-about 1909.

Table A-2.—Experimental operations involving leach solution application—Continued

Operation	Mountain City mine	Van Dyke deposit
Location	Mountain City, NV.	Miami, AZ.
Host rock	Phyllite and quartzite lenses in shale.	Pinal Schist.
Copper minerals:		
Principal	Chalcocite.	Chrysocolla.
Minor	Chalcopyrite, cuprite, malachite, chrysocolla, azurite, native.	Azurite, malachite.
Average ore grade ... pct	0.93-1.10	0.5
Ore preparation	Blockcaving.	Wells drilled and hydrofraced.
Area involved ... ft²	54,400	1st test involved 2 wells 75 ft apart, 2d test 10,000 (5-spot pattern).
Maximum depth ... ft	400	1st test 1,000, 2d test 1,200.
Quantity of material ... T	793,000	Not available.
Average fragment size ... in	Not available.	Do.
Leach solution	Dilute H_2SO_4.	Dilute H_2SO_4.
pH	Not available.	Not available.
Solution application	Injection wells (25-ft centers, 150-200 ft deep, 2-in-diam PVC casing).	Injection well (4-in diam).
Flow rate ... gpm	Grossly variable, 0-25 per hole.	Not available (1:1 ratio between injection and production).
Solution recovery	Drift on 400-ft level that drained to mine sump on 450-ft level.	4-in-diam recovery well (1 for 1st test, 4 for 2d test).
Water table	Above water table.	Below water table.
Pregnant solution copper content ... gpL	0.5-0.6	Not available.
Type of copper recovery	Precipitation on scrap iron.	SX-EW (neighboring plants).
Consumption, lb/lb Cu:		
Acid (H_2SO_4)	Not available.	Not available.
Iron	Do.	Not applicable.
Copper production ... lb/d	4,800 (average).	Not available.
Employees	Not available.	Do.
Active dates	March 1974-December 1974.	January 1976-October 1980.

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